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PILOT-PLANT INVESTIGATION OF COPPER-NICKEL ORE FROM GREAT LAKES NICKEL CORPORATION LIMITED, PARDEE TOWNSHIP, THUNDER BAY DISTRICT, ONTARIO

by

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MINERAL PROCESSING DIVISION

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SUMMARY OF RESULTS

Typical bulk flotation results obtained for this low-grade copper-nickel ore were as follows:

	Wt %	Cu %	Ni %	Pt oz/ton	Pd oz/ton
Bulk concentrate	4-5	10-13	4-5.5	0.15	0.7 -0.8
Tailing		0.03-0.04	0.05-0.08	0.003	0.002-0.005
Feed		0.5 -0.6	0.25-0.30	0.01	0.035

Recovery in concentrate %

Copper	93-94
Nickel	70-80
Platinum	67
Palladium	87

Generally, the lower tailing losses were accompanied by the production of a lower grade of concentrate.

The fineness of grind which was varied from 50% to 68% -200 mesh did not appear to have any effect on results.

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Typical results achieved by employing selective flotation to produce separate copper and nickel concentrates are given below:

		Assays				Distribution			
	Wt %	Cu	Ni	Pt	Pd	Cu	Ni	Pt	Pd
Copper conc	1.6	24.00	1.28	0.048	0.42	79.5	8.5	12.0	26.5
Nickel conc	3.0	2.00	5.10	0.092	0.46	12.5	63.7	43.2	54.7
Tailing	95.4	0.04	0.07	0.003	0.005	8.0	27.8	44.8	18.8

In one test run a copper concentrate containing only 0.64% nickel was produced but this result could not be duplicated.

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INTRODUCTION

Location of Property

The property is located in Pardee Township about 40 miles southwest of Fort William, Ontario.

Shipment

A total of 112 tons of ore was shipped from the property as follows:

Lot No.	Weight, Tons	Date Received
1	61	February 14, 1967
2	9	April 10, 1967
3	42	April 24, 1967

In addition, two small samples of lump ore weighing 112 lb and 31 lb were received on December 5, 1966 and January 26, 1967 respectively. These were used for bench-scale tests.

Purpose of Investigation

In a letter dated March 4, 1966, Mr. Walter H. Hood, General Manager of Great Lakes Nickel Corporation Limited, requested that a pilot-plant investigation be carried out on the ore in order to develop a suitable flowsheet for a large-scale plant. In subsequent discussions with Dr. T. Salman, Mineral Dressing Consultant for Great Lakes Nickel, it was agreed that the primary object of the investigation would be to establish optimum conditions for the flotation of a bulk sulphide concentrate. When this had been accomplished, copper-nickel selective flotation would be tried with the aim of producing a copper concentrate assaying less than 1% nickel along with a nickel concentrate in which the nickel content would be at least 3 times the copper content.

Mineralogy of the Ore and Chemical Analysis

Several mineralized hand specimens from the 112-lb shipment were submitted to the Mineral Sciences Division for mineralogical examination*. The results of the examination were as follows:

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*From Internal Report MS 67-6 by Dr. L. J. Cabri

The mineralized rock is a fresh, medium to coarse-grained olivine-gabbro. The non-metallic gangue consists mostly of feldspar, olivine, clinopyroxene, and minor hornblende.

The metallic minerals are pentlandite, cubanite, pyrrhotite, chalcopyrite, magnetite and ilmenite. Two small grains of maucherite (nickel arsenide) were found within cubanite in one polished section. A lighter coloured unidentified second phase occurs in the maucherite.

The pentlandite is generally fractured and replaced by numerous fine veins of pyrrhotite and magnetite. It also occurs as tiny remnant "trains" in pyrrhotite. The cubanite occurs in larger quantities than chalcopyrite in some sections and appears to be more closely related to the pentlandite than the chalcopyrite. There are lamellar intergrowths of cubanite with chalcopyrite, and lamellae of chalcopyrite in pentlandite. Lamellae of pyrrhotite are also common in cubanite.

Since it has been stated that the mineral characteristics of the Great Lakes orebody are uniform throughout its extent, these results would apply with equal validity to the ore tested in the pilot-plant.

TABLE 1

Head Sample Analysis

Ore Sample	%				oz./ton			
	Cu	Ni	Fe	S	Pt	Pd	Au	Ag
Lot 1	0.51	0.26	6.77	0.95	0.01	0.033	trace	0.04
Lot 2	0.60	0.29						
Lot 3	0.44	0.23	6.95	0.94	0.01	0.03		
112 lb shipment	0.88	0.41	7.46	1.69	0.02	0.047	0.006	0.238
31 lb shipment	0.95	0.45						

Bench-Scale Tests

Lab investigations of the ore had previously been carried out by Dr. T. Salman and also by INCO. Since this work showed that the sulphides floated readily to give a satisfactory bulk concentrate grade with good recoveries, it was not necessary to conduct a comprehensive lab investigation at the Mines Branch. However, in order to become familiar with the flotation characteristics of the ore and to establish an optimum grind, a number of bench-scale tests employing bulk flotation were done on the samples submitted for this purpose. A few selective flotation tests were also tried. The results were promising and indicated that it might be possible to produce copper and nickel concentrates approaching the required specifications.

General Procedure

All pilot-plant test runs were conducted during regular working hours and were generally of 6 hours duration. Samples of various products were taken once every 1/2 hour during the last 1 1/2 hours of the test run when it was assumed that conditions had stabilized. Pilot-plant feed rate was 750 lb /hour except for the first two test runs in which a 500 lb/ hour feed rate was employed. Sample preparation and analysis were done by Mines Branch personnel on an overtime basis following the completion of each test run.

All concentrates produced in the pilot-plant were saved, filtered and stored in drums.

Dr. T. Salman visited the Mines Branch on a number of occasions during the course of the investigation to discuss results and the direction of future testing.

DETAILS OF INVESTIGATION

Crushing

Each lot of ore was initially crushed to 1 1/2 inches. During the course of crushing, a head sample amounting to about 2% of the weight of the ore was taken continuously by a Synder automatic sampler. The stockpile of 1 1/2-inch ore was thoroughly mixed by repiling it several times with a front end loader. Small quantities of ore were then drawn from the stockpile and crushed further to -1/4 inch as required to provide feed for the pilot-plant.

Grinding and Classification

Single-stage, closed-circuit grinding was employed. A 30 x 48-in. ball mill was used along with a 30-in.- diameter Sweco vibrating screen as a classifier. In 3 test runs the Sweco separator was replaced by a 14 x 8-in. Dorr rake classifier.

Bulk Flotation

A total of 19 test runs was carried out in which the copper and nickel sulphides were floated into a single bulk concentrate. Variables investigated were as follows:

(1) The use of copper sulphate

(2) Fineness of grind

(3) Mechanical classifier vs vibrating screen

(4) Flotation contact time

The only problem encountered in bulk flotation was the difficulty in maintaining high concentrate grades due to the excessive flotation of gangue minerals. The following schemes were tried in attempts to improve concentrate grades:

(1) Regrinding of rougher concentrate followed by thickening reground product before cleaning - Guartec SJ/M was added to the thickener to serve as a settling agent and gangue depressant.

(2) Same scheme as above but Separan NP 10 substituted for Guartec – because of the small amount of Separan used (0.007 lb /ton) it was assumed that it had no effect on flotation and served as a settling agent only.

(3) Reground cleaner tailing before recirculation.

(4) Sodium silicate was added to ball mill.

(5) Sodium silicate was added to cleaners.

Results for bulk flotation along with the variables tested are given in Tables 3 and 5. In Table 3 all test runs were done on Lot 1 ore and Test Run 7 was the standard or comparison test run from which procedure was varied. Flowsheet for this test run is illustrated in Figure 1 while reagents and conditions are given in Table 2.

In Table 5 Test Run 30 was the standard test run. All the test runs in this series were done on Lot 3 ore and differed mainly from those in Table 3 in that a longer flotation contact time was employed. Reagents and conditions for the standard test run are given in Table 4 while the flowsheet is illustrated in Figure 2.

TABLE 2

Reagents and Conditions for Test Run 7 (Standard Test Run, Lot 1)

Operation	Point of Reagent Addition	Reagents, lb /ton	
		Z-6*	Dowfroth 250
Conditioning	Feed		0.016
Bulk rougher flotation	No. 1 Cell		0.004
	No. 2 Cell	0.011	0.014
	No. 3 Cell	0.011	
	No. 4 Cell		0.004
	No. 5 Cell	0.032	

pH			% Solids		
	Rougher flotation feed	9.6		Ball mill discharge	68
	Cleaner tailing	9.2		Rougher flotation Feed	28
				Cleaner tailing	< 5

*Potassium amyl xanthate

TABLE 3

Bulk Flotation Results on Lot 1 Ore

Test Run No.	Test Variables	Grind % -200m	Product	Wt %	Assays %*				Distribution %	
					Cu	Ni	Insol	S	Cu	Ni
7	Standard test run	68.0	Bulk cleaner conc	4.0	12.88	5.50	22.30		93.0	79.1
			Tailing	96.0	0.04	0.06			7.0	20.9
			Feed (calcd)	100.0	0.55	0.28			100.0	100.0
6	Coarser grind	57.2	Bulk cleaner conc	4.2	10.66	4.79	23.26		93.1	80.8
			Tailing	95.8	0.035	0.05		0.15	6.9	19.2
			Feed (calcd)	100.0	0.48	0.25			100.0	100.0
8	Copper sulphate to conditioner, 0.20 lb/ton	66.6	Bulk cleaner conc	3.4	14.56	5.50	17.56		92.1	74.7
			Tailing	96.6	0.044	0.066		0.18	7.9	25.3
			Feed (calcd)	100.0	0.54	0.25			100.0	100.0
9	Reground rougher conc to 90% -325m and thickened before cleaning, added Guartec (0.028 lb/ton) to thickener	62.6	Bulk cleaner conc	3.1	15.56	6.00	13.26		88.4	73.0
			Tailing	96.9	0.066	0.072		0.20	11.6	27.0
			Feed (calcd)	100.0	0.55	0.26			100.0	100.0
10	As in Test Run 9 but increased Gunrtec feed rate to thickener to 0.062 lb/ton	62.9	Bulk cleaner conc	2.4	17.60	8.75	9.30		83.0	70.9
			Tailing	97.6	0.09	0.09		0.23	17.0	29.1
			Feed (calcd)	100.0	0.52	0.30			100.0	100.0
11	As in Test Runs 9 and 10 but replaced Guartec with Separan (0.007 lb/ton)	64.3	Bulk cleaner conc	3.0	16.84	6.00	13.20		87.7	68.4
			Tailing	97.0	0.073	0.086		0.21	12.3	31.6
			Feed (calcd)	100.0	0.58	0.26			100.0	100.0
12	Reground cleaner tailing to 91% -325m before recirculation	62.5	Bulk cleaner conc	3.0	16.00	6.00	14.26		90.9	67.4
			Tailing	97.0	0.06	0.09		0.22	9.1	32.6
			Feed (calcd)	100.0	0.53	0.27			100.0	100.0

*From Internal Reports MS-AC-157, 171, 173, 177, 183, 192 and 194.

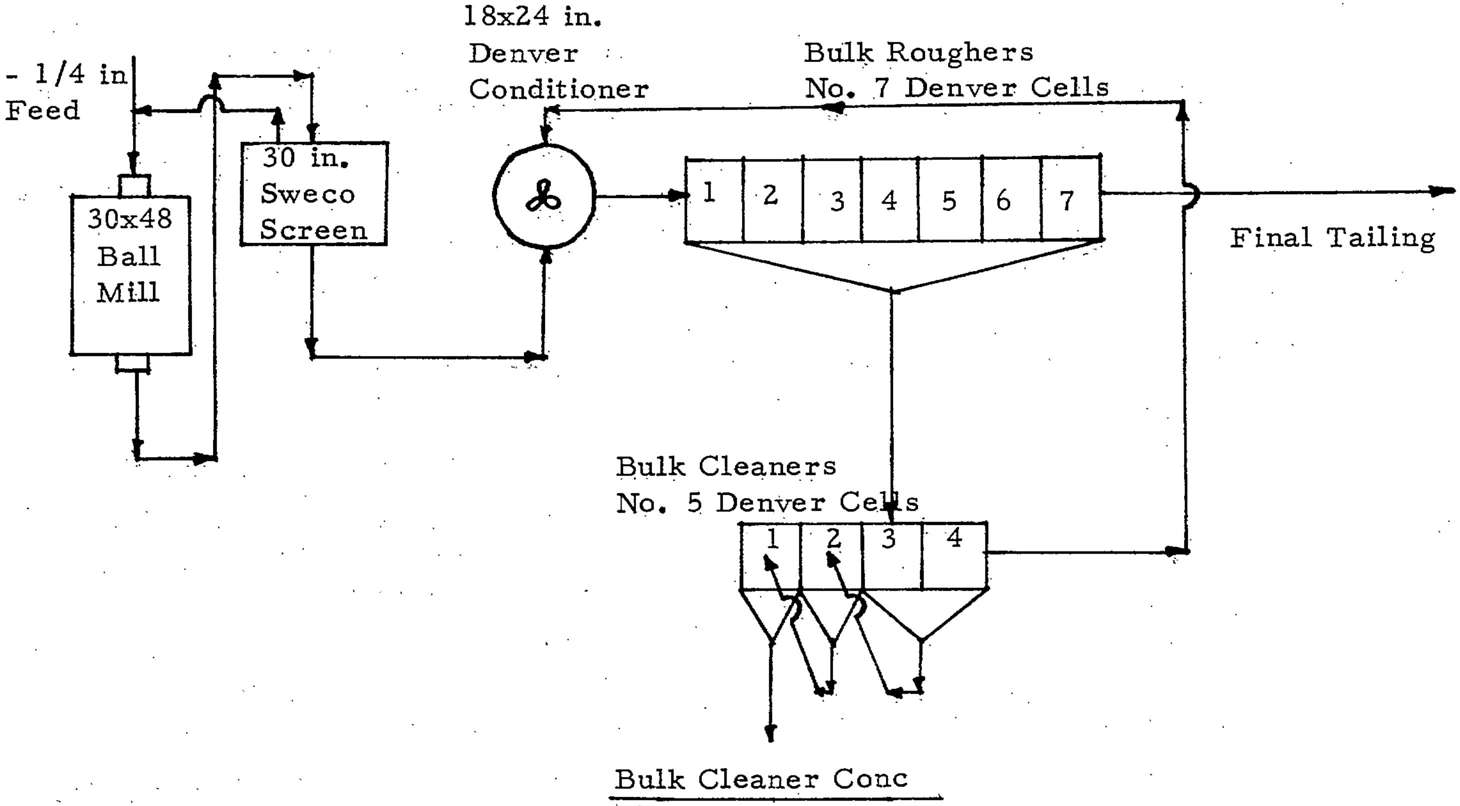

Figure 1. Flowsheet for Bulk Flotation of Lot 1 Ore

TABLE 4

Reagents and Conditions for Test Run 30 (Standard Test Run, Lot 3)

Operation	Point of Reagent Addition	Reagents, lb/ton	
		Z-6	Dowfroth 250
Conditioning	Feed		0.018
Bulk rougher flotation	No. 1 cell		0.011
	No. 2 cell	0.011	0.014
	No. 3 cell	0.011	
	No. 4 cell		0.007
Bulk scavenger flotation	No. 1 cell	0.053	
	No. 3 cell	0.025	0.021
pH Rougher flotation feed 8.8	% Solids	Ball mill discharge 64	
		Conditioner feed 38	

TABLE 5

Bulk Flotation Results on Lot 3 Ore

Test Run No.	Test Variables	Grind % -200	Product	Wt %	Assays %*				Distribution %	
					Cu	Ni	Insol	S	Cu	Ni
30	Standard test run	55.2	Bulk cleaner conc	4.7	10.60	4.38	24.50		94.6	78.3
			Tailing	95.3	0.03	0.06		0.17	5.4	21.7
			Feed (calcd)	100.0	0.53	0.26			100.0	100.0
			No. 4 Rghr cell tailing		0.06	0.09				
33	Mechanical classifier in place of vibrating screen	60.6	Bulk cleaner conc	3.8	13.40	5.00	19.26		94.6	71.2
			Tailing	96.2	0.03	0.08		0.23	5.4	28.8
			Feed (calcd)	100.0	0.54	0.27			100.0	100.0
			No. 4 Rghr cell tailing		0.08	0.11				
34	As in Test Run 33 but coarser grind	55.7	Bulk cleaner conc	4.9	10.00	3.82	27.26		94.1	76.4
			Tailing	95.1	0.032	0.06		0.19	5.9	23.6
			Feed (calcd)	100.0	0.52	0.24			100.0	100.0
			No. 4 Rghr cell tailing		0.059	0.093				
31	Sodium silicate to ball mill, 1.06 lb/ton	52.7	Bulk cleaner conc	3.4	13.78	5.00	17.34		94.2	69.0
			Tailing	96.6	0.03	0.08		0.22	5.8	31.0
			Feed (calcd)	100.0	0.50	0.25			100.0	100.0
			No. 4 Rghr cell tailing		0.06	0.12				
46	Coarser grind	50.3	Bulk cleaner conc	3.8	13.20	4.50	19.26		92.8	68.7
			Tailing	96.2	0.04	0.08		0.24	7.2	31.3
			Feed (calcd)	100.0	0.53	0.25			100.0	100.0
			No. 4 Rghr cell tailing		0.08	0.10				
45	Same grind as Test Run 46 but with sodium silicate (0.52 lb/ton) to cleaners	49.6	Bulk cleaner conc	5.0	10.40	4.00	25.40		94.8	74.9
			Tailing	95.0	0.03	0.07		0.23	5.2	25.1
			Feed (calcd)	100.0	0.55	0.27			100.0	100.0
			No. 4 Rghr cell tailing		0.05	0.08				

*From Internal Reports MS-AC-67-382, 386, 391, 392 and 508.

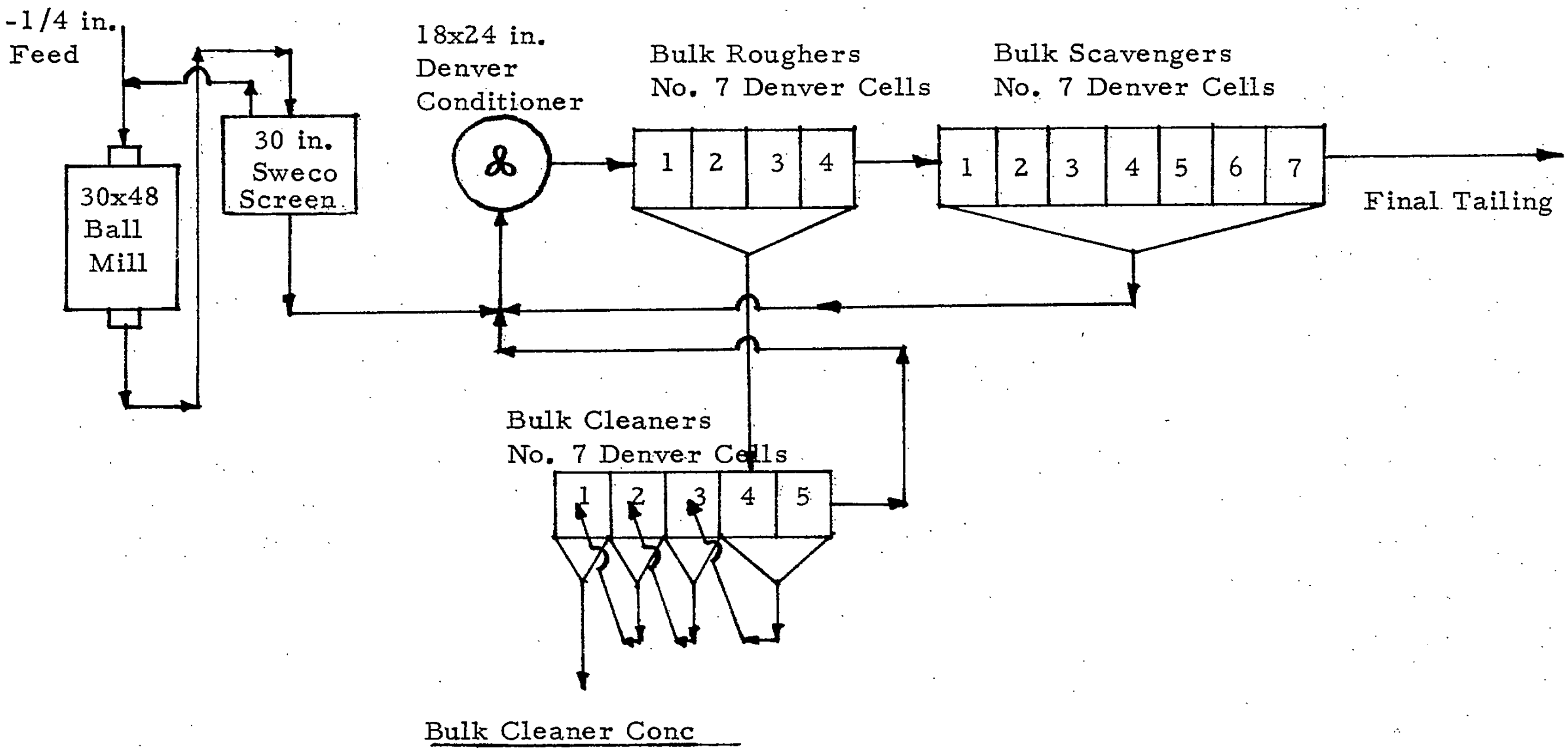

Figure 2. Flowsheet for Bulk Flotation of Lot 3 Ore

Recovery of Platinum-Group Metals

Three sets of bulk flotation products were assayed for platinum group metals and recoveries calculated. Results are shown in Table 6.

TABLE 6

Recovery of Platinum-Group Metals by Bulk Flotation

Test Run No.	Product	Wt %	Assays* Cu	Ni	Pt	Pd	Rh	Au	Ag	Distribution % Cu	Ni	Pt	Pd	Ag
6	Bulk cl conc	4.2	10.66	4.79	0.33	0.82		0.06	1.99	93.1	80.8	87.9	94.7	62.3
	Tailing	95.8	0.035	0.05	0.002	0.002		tr	0.053	6.9	19.2	12.1	5.3	37.7
	Feed (calcd)	100.0	0.48	0.25	0.016	0.037			0.135	100.0	100.0	100.0	100.0	100.0
	Heads (assay)		0.46	0.26	0.011	0.038		tr	0.101					
Comp**	Bulk Cl conc	4.1	12.64	4.95	0.15	0.77	0.012	0.055	2.32	92.6	72.4	68.0	89.1	74.9
	Tailing	95.9	0.043	0.08	0.003	0.004		tr	0.033	7.4	27.6	32.0	10.9	25.1
	Feed (calcd)	100.0	0.56	0.28	0.009	0.035			0.126	100.0	100.0	100.0	100.0	100.0
	Heads (assay)		0.55	0.28	0.008	0.035		0.004	0.101					
46	Bulk cl conc	3.8	13.20	4.50	0.15	0.70	0.016			92.8	68.7	66.1	84.5	
	Tailing	96.2	0.04	0.08	0.003	0.005				7.2	31.3	33.9	15.5	
	Feed (calcd)	100.0	0.53	0.25	0.009	0.031				100.0	100.0	100.0	100.0	
	Heads (assay)		0.52	0.25	0.008	0.033								

* From Internal Reports 67-157, 196, 363, 508 and 568.
**Weighted composite of Test Runs 1, 2, 5, 6, 7 and 8.

Mineralogical Examination of Bulk Cleaner Concentrate

A sample of bulk cleaner concentrate produced in Test Run 7 was screened and the screen fractions were sent to the Mineral Sciences Division for mineralogical examination. It was requested that the gangue minerals be identified and that the degree of liberation of the copper and nickel minerals be established.

Table 7 gives the distribution of copper, nickel and insolubles in the various sizes.

TABLE 7

Distribution of Copper, Nickel and Insolubles in Various Size Fractions of Bulk Cleaner Concentrate (Test Run 7)

Size Fraction	Wt %	Assays %*			Distribution %		
		Cu	Ni	Insol	Cu	Ni	Insol
+100 m	1.6	12.88	2.69	31.18	1.6	0.8	2.2
+200 m	23.0	12.00	4.50	27.48	22.0	19.5	27.7
+325 m	16.8	12.00	6.00	21.60	16.1	19.0	15.9
-325 m	58.6	12.88	5.50	21.16	60.3	60.7	54.2
Total	100.0	12.53	5.31	22.85	100.0	100.0	100.0

*From Internal Report 67-171

The results of the mineralogical examination* were as follows:

The ore minerals in the size fractions consist mainly of chalcopyrite, cubanite, pentlandite and pyrrhotite. A few grains each of ilmenite, magnetite and covellite were also found.

The chalcopyrite and cubanite grains are essentially free from the other sulphides and gangue minerals, although they are sometimes intergrown with each other. It is estimated that the degree of liberation of the chalcopyrite and cubanite from the other minerals in the +100, +200, +325 and -325 mesh size fractions is approximately 80, 85, 90 and greater than 95% respectively. The grains of cubanite and chalcopyrite that are not free occur as combined grains with pyrrhotite, pentlandite and rarely ilmenite, and as small inclusions in the gangue.

*From Internal Report MS 67-34 by D. Owens

The degree of freedom of the pentlandite is similar to that of the copper-bearing minerals.

The gangue is composed chiefly of feldspar and orthopyroxene, with small amounts of clino-pyroxene, biotite and chlorite. The proportions of these minerals in all four size fractions are approximately the same and it can be assumed that the insoluble content of the concentrate is due largely to the presence of the feldspar and orthopyroxene. Most of the gangue minerals are free from sulphide inclusions, and it appears that the gangue minerals in the size fractions are mainly due to entrapment.

Distribution of Metal Losses in Various Size Fractions of Tailing

Several tailing samples were sized and the size fractions assayed in order to determine metal distribution. Results for tailings from Test Runs 12 and 31 are given in Table 8.

TABLE 8

Distribution of Metal Losses in Various Size Fractions of Tailing

Test Run No.	Size Fraction	Wt %	Assays %* Cu	Assays %* Ni	Distribution % Cu	Distribution % Ni
12	+100 m	7.2	0.05	0.08	5.1	6.3
	+200 m	31.3	0.03	0.06	13.3	20.5
	+325 m	17.1	0.03	0.06	7.3	11.2
	+400 m	3.8	0.10	0.11	5.4	4.6
	-400 m	40.6	0.12	0.13	68.9	57.4
	Tailing (calcd)	100.0	0.07	0.09	100.0	100.0
	Tailing (assay)		0.06	0.09		
31	+65 m	3.3	0.06	0.08	6.8	4.8
	+100 m	16.8	0.04	0.06	23.2	18.3
	+150 m	18.0	0.03	0.05	18.7	16.4
	+200 m	11.9	0.02	0.05	8.2	10.8
	+56 mu	5.1	0.02	0.07	3.5	6.5
	+40 mu	12.7	0.02	0.04	8.8	9.2
	+28 mu	9.1	0.01	0.04	3.1	6.6
	+20 mu	6.8	0.02	0.04	4.7	4.9
	+14 mu	4.6	0.02	0.05	3.2	4.2
	+10 mu	3.3	0.02	0.05	2.3	3.0
	-10 mu	8.4	0.06	0.10	17.5	15.3
	Tailing (calcd)	100.0	0.03	0.06	100.0	100.0
	Tailing (assay)		0.03	0.08		

*From Internal Reports MS-AC-67-194 and 482.

Metal Distribution in Size Fraction of
Classifier Overflow and Screen Undersize

In order to determine to what extent differential classification of sulphides occurred when the mechanical classifier was used in place of the vibrating screen, samples of classifier overflow and screen undersize with the same percentage passing 200 mesh were sized and the size fractions assayed for copper and nickel. Results are given in Table 9.

TABLE 9

Distribution of Copper and Nickel in Various Size Fractions of Classifier Overflow and Screen Undersize

Product	Size Fraction	Wt %	Assays %*		Distribution %	
			Cu	Ni	Cu	Ni
Screen Undersize, Test Run 30	+65 m	2.2	0.58	0.20	2.3	1.7
	+100 m	12.0	0.34	0.15	7.4	6.8
	+150 m	17.0	0.40	0.19	12.3	12.2
	+200 m	12.7	0.49	0.22	11.3	10.6
	+325 m	13.9	0.53	0.28	13.3	14.8
	+400 m	4.0	0.58	0.31	4.2	4.7
	-400 m	38.2	0.71	0.34	49.2	49.2
	Feed (calcd)	100.0	0.55	0.26	100.0	100.0
	Feed (assay)		0.56	0.25		
	-200 m	56.1			67.7	68.7
Classifier Overflow, Test Run 34	+65 m	5.2	0.15	0.06	1.5	1.4
	+100 m	10.3	0.17	0.08	3.5	3.7
	+150 m	15.7	0.29	0.10	9.0	7.1
	+200 m	13.0	0.40	0.16	10.3	9.4
	+325 m	14.1	0.51	0.22	14.2	14.0
	+400 m	4.3	0.60	0.27	5.1	5.2
	-400 m	37.3	0.76	0.35	56.4	59.2
	Feed (calcd)	100.0	0.50	0.22	100.0	100.0
	Feed (assay)		0.52	0.24		
	-200 m	55.8			75.7	78.4

*From Internal Reports MS-AC-67-472 and 482.

Determination of Work Index of Ore

The Bond work index of the ore was determined by the comparative method employed at the Mines Branch.* In this method a sample of ore is ground in a lab rod mill using identical procedure employed for grinding a reference ore whose work index is known. Since the work required to grind both ores is equivalent, the unknown work index can readily be calculated. The work index calculated for a sample of Lot 1 ore was 20.3 at a grind of 53.1% -200 mesh.

* "A Simple Method of Determining the Grindability of Ores" by T.F. Berry and R.W. Bruce, Canadian Mining Journal, July, 1966.

Copper-Nickel Selective Flotation

Description of Methods Employed

Three methods were tried to produce separate copper and nickel concentrates as follows:

Method I

(1) The easily floatable copper minerals were floated off and the copper rougher concentrate was cleaned at a high pH with lime. The aim of this first treatment step was to reduce the copper to nickel ratio in the ore with the hope of improving the efficiency of the subsequent copper-nickel separation step. Initially, only frother was used to float copper but this was found to be inadequate and it was found necessary to add Z-200, a selective copper promoter.

(2) Amyl xanthate and frother were then added to recover the remaining copper and nickel sulphides in a copper-nickel concentrate which was cleaned without additional reagents.

(3) A copper-nickel separation was made on the copper-nickel cleaner concentrate by floating off a copper concentrate after first depressing the nickel minerals with high lime alkalinity. The copper concentrate from this operation, after being upgraded by multi-stage cleaning, could then be combined with the copper concentrate produced in step (1) to form the final product for shipment to a copper smelter.

Method II

(1) A copper concentrate was floated selectively from the ore by using high lime alkalinity to depress the nickel minerals and Z-200 as the copper promoter. The copper concentrate was upgraded by multi-stage cleaning at a high pH with lime.

(2) Sulphuric acid was added to lower the pH of the copper rougher tailing and after activation with copper sulphate a nickel concentrate was floated off with amyl xanthate and frother. The nickel concentrate was upgraded by multi-stage cleaning without additional reagents.

Method III

This method was similar to Method II except that sodium sulphite was substituted for lime as a depressant for the nickel minerals during the copper rougher float and Aerofloat 238 was substituted for Z-200 as the copper promoter. Also, since copper rougher flotation was carried out at a low pH it was not necessary to employ sulphuric acid for subsequent nickel flotation.

Method I Tests

Five Test Runs (No. 13 to 17), all on Lot 1 ore, were carried out using this method. In all test runs nickel depression was unsatisfactory, resulting in copper concentrates containing much higher amounts of nickel than the specified content of 1% or less. The best results were obtained in Test Run 15. The flowsheet employed for this test is illustrated in Figure 3 while reagents and conditions and results are given in Tables 10 and 11 respectively.

TABLE 10

Reagents and Conditions for Method I Test Run 15

Operation	Point of Reagent Addition	Reagents, lb/ton			
		Z-200	Z-6	DF 250	Lime
Conditioning	Feed	0.027			
Copper rougher flotation	No. 1 Cell			0.005	
Cu-Ni rougher flotation	No. 2 Cell		0.011	0.011	
	No. 3 Cell		0.014		
	No. 4 Cell			0.007	
	No. 5 Cell		0.032		
Copper-nickel separation	Feed				0.57
	No. 2 Cell				0.39
Copper cleaning	No. 2 Cell				0.21

	pH
Copper rougher flotation feed	9.6
Copper cleaner cell No. 1	11.5
Copper cleaner cell No. 2	12.3
Copper cleaner cell No. 3	11.9
Copper cleaner cell No. 4	11.2
Cu-Ni separation feed	12.4
Cu-Ni separation cell No. 1	9.4
Cu-Ni separation cell No. 2	10.5
Cu-Ni separation cell No. 3	11.1
Cu-Ni separation cell No. 4	11.4
Cu-Ni separation cell No. 5	12.4

	% Solids
Ball mill discharge	66
Conditioner feed	41

Primary grind (screen undersize), 61.0% -200 mesh

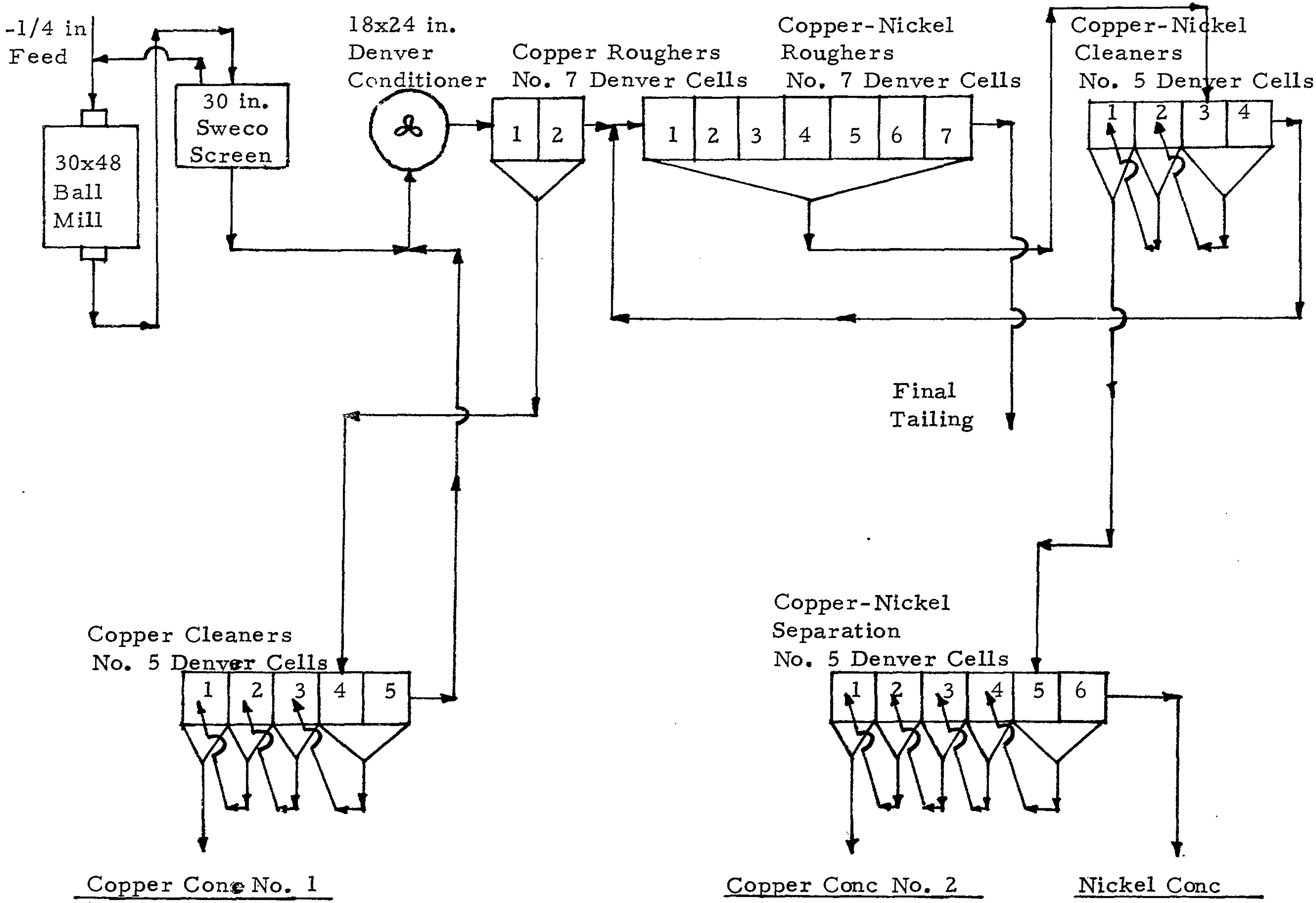

Figure 3. Flowsheet for Selective Flotation Method I

TABLE 11

Results of Method I Test Run 15

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate No. 1	1.56	25.76	2.50	2.98		71.7	16.4
Copper concentrate No. 2	0.27	20.80	6.00	3.16		10.0	6.8
Nickel concentrate	2.30	2.96	5.00	40.55		12.1	48.5
Final tailing	95.87	0.036	0.07		0.16	6.2	28.3
Feed (calcd)	100.00	0.56	0.24			100.0	100.0
Heads (assay)		0.56	0.26				
Cu conc No. 1 and No. 2	1.83	25.03	3.02			81.7	23.2
Copper rougher concentrate		17.76	4.50				
Copper rougher tailing		0.17	0.21				
Copper cleaner tailing		11.72	5.29				
Copper-nickel rougher conc		2.60	3.00				
Copper-nickel cleaner conc		4.85	6.00				

*From Internal Report MS-AC-67-210.

Method II Tests

This method was more thoroughly investigated than Method I. A total of 11 test runs was carried out, 7 on Lot 1 (No. 19 to 25) and 4 on Lot 2 (No. 26 to 29). After many attempts, a copper concentrate assaying less than 1% nickel was finally produced in Test Run 27. The changes made from the initial test procedure in order to achieve these results were as follows:

(1) The addition of higher amounts of lime to both copper roughers and cleaners.

(2) The addition of sodium silicate to the ball mill for gangue depression in the copper roughers.

(3) Regrind of copper rougher concentrate before cleaning.

The flowsheet employed for Test Run 27 is shown in Figure 4. Reagents and conditions and results are given in Tables 12 and 13 respectively.

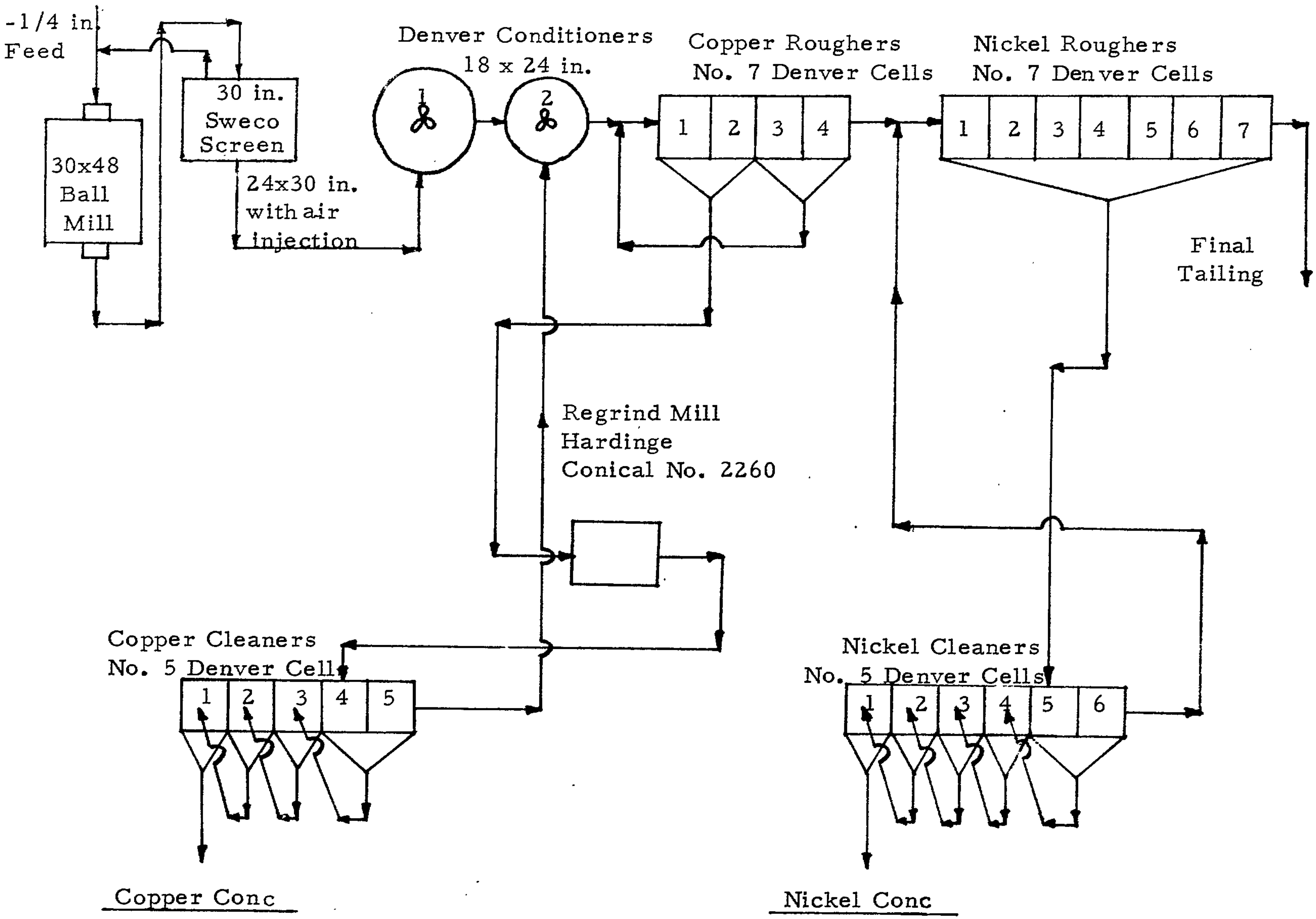

Figure 4. Flowsheet for Selective Flotation Method II

TABLE 12

Reagents and Conditions for Method II Test Run 27

Operation	Point of Reagent Addition	Reagents, lb/ton						
		Z-200	Z-6	DF 250	Lime	Sod. Silicate	$CuSO_4$	H_2SO_4
Grinding	Feed	0.021			6.35	1.4		
Cu rougher flotation	No. 1 Cell			0.005				
Ni rougher flotation	No. 1 Cell						0.21	8.44
	No. 2 Cell		0.035	0.057				
	No. 3 Cell		0.032					
	No. 4 Cell			0.021				
	No. 5 Cell		0.042					
Regrinding	Feed				0.85			
Copper cleaning	No. 1 Cell				0.16			
	No. 2 Cell				0.21			
Nickel cleaning	No. 4 Cell			0.014				

	pH
Copper rougher flotation feed	12.0
Nickel rougher cell No. 1	7.9
Copper cleaner feed	12.3
Copper cleaner cell No. 1	11.9
Copper cleaner cell No. 2	11.9
Copper cleaner cell No. 3	11.7
Copper cleaner cell No. 4	12.0

	% Solids
Ball mill discharge	62
Conditioner feed	40

Screen Analysis	
Primary grind (screen u' size)	54.3% -200 mesh
Regrind feed	70.3% -325 mesh
Regrind discharge	83.3% -325 mesh

TABLE 13

Results of Method II Test Run 27

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.75	29.36	0.82	0.64		76.6	5.0
Nickel concentrate	1.65	6.00	9.12	20.70		14.8	51.8
Final tailing	96.60	0.06	0.13		0.22	8.6	43.2
Feed (calcd)	100.00	0.67	0.29			100.0	100.0
Heads (assay)		0.67	0.29				
Copper rougher concentrate		23.20	1.65				
Copper rougher tailing		0.20	0.27				
Copper cleaner tailing		21.20	1.65				
Nickel rougher concentrate		4.20	5.50				
Nickel cleaner tailing		2.08	2.12				

*From Internal Report MS-AC-67-356.

In Test Runs 28 and 29, Test Run 27 procedure was followed in attempts to duplicate the good results obtained, but with changes designed to improve recoveries. These were as follows:

Test Run 28

(1) Put concentrate from copper rougher cells 3 and 4 to regrind mill.

(2) Cut nickel cleaning from 5 stages to 3 stages.

Test Run 29

(1) Replaced 170 lb of ceramic balls in regrind mill with 350 lb of steel balls in order to obtain a finer grind.

(2) Added 3 No. 7 Denver cells to nickel roughers. Froth from these cells was recirculated to the head of the nickel roughers.

Results of Test Runs 28 and 29 are given in Tables 14 and 15 respectively.

TABLE 14

Results of Method II Test Run 28

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.80	27.60	1.50	1.10		80.0	9.3
Nickel concentrate	1.79	4.80	9.34	24.90		13.8	57.5
Final tailing	96.41	0.04	0.10		0.18	6.2	33.2
Feed (calcd)	100.00	0.62	0.29			100.0	100.0
Heads (assay)		0.62	0.29				
Copper rougher concentrate		19.20	1.85				
Copper rougher tailing		0.14	0.28				
Copper cleaner tailing		15.64	1.90				
Nickel rougher concentrate		2.73	5.75				
Nickel cleaner tailing		1.15	1.85				

*From Internal Report MS-AC-67-359.

TABLE 15

Results of Method II Test Run 29

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.92	26.40	1.61	2.00		84.7	10.7
Nickel concentrate	1.89	2.82	8.60	30.36		8.9	56.1
Final tailing	96.19	0.04	0.10		0.14	6.4	33.2
Feed (calcd)	100.00	0.60	0.29			100.0	100.0
Heads (assay)		0.60	0.29				
Copper rougher concentrate		18.40	2.00				
Copper rougher tailing		0.11	0.26				
Copper cleaner tailing		12.00	2.15				
Nickel rougher concentrate		1.76	4.32				
Nickel cleaner tailing		1.00	2.25				

*From Internal Report MS-AC-67-365.

Method III Tests

Ten test runs were carried out using this method (No. 35 to 44) all on Lot 3 ore. A primary grind of approximately 52% -200 mesh was employed in all test runs. The amount of lime fed to the regrind mill and copper cleaners was much higher than that employed for Method II. Variables investigated were as follows:

(1) A high pulp temperature in copper cleaning.

(2) The retention of the copper cleaner tailing as a separate product for mixing with the nickel concentrate.

In the nickel circuit several schemes were tried in attempts to lower the gangue content of the nickel concentrate as follows:

(1) The addition of copper sulphate and soda ash to the cleaners to obtain a more heavily mineralized type of froth from which gangue particles would be crowded out.

(2) Thickening of the nickel rougher concentrate before cleaning along with the addition of copper sulphate, soda ash and amyl xanthate to the cleaners for the same reason as (1).

(3) The addition of sodium silicate to the cleaners.

In the first 2 test runs it was found difficult to maintain a high grade of copper rougher concentrate due to the flotation of excessive amounts of gangue minerals. The high gangue content was undesirable as most of it was subsequently rejected to the copper cleaner tailing which, at this point in the investigation, was being saved as a separate product. In order to upgrade the copper rougher concentrate before it was sent to the main cleaning operation, 2 stages of precleaning were cut into the circuit starting with Test Run 37. No reagents were added to the copper precleaners and the precleaner tailing was recirculated to the head of the copper roughers.

The best set of results were obtained for Test Run 42. In this test run a high pulp temperature was used in copper cleaning and was obtained by replacing the cold water normally used for the launder sprays with hot water. Also scheme (2) described above was used for gangue depression in the nickel cleaners. The flowsheet used for Test Run 42 is illustrated in Figure 5 while reagents and conditions and results are given in Tables 16 and 17 respectively.

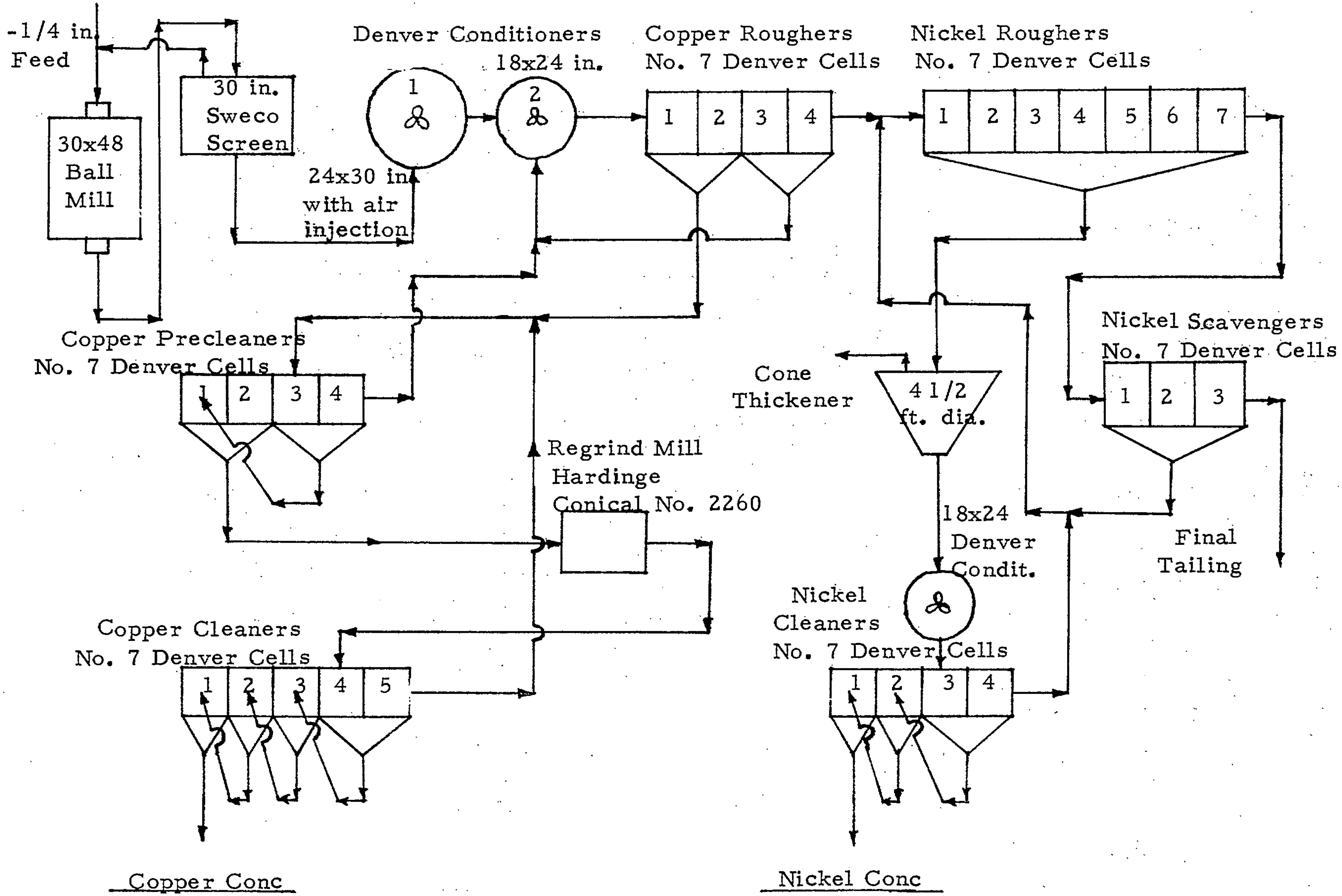

Figure 5. Flowsheet for Selective Flotation Method III

TABLE 16

Reagents and Conditions for Method III Test Run 42

Operation	Point of Reagent Addition	Reagents, lb/ton						
		Aero 238	Z-6	DF 250	Lime	Na_2CO_3	Na_2SO_3	$CuSO_4$
Grinding	Feed					1.06	1.06	
Conditioning	Feed of No. 2			0.011				
Cu rougher flotation	No. 1 Cell	0.018						
	No. 2 Cell			0.014				
	No. 3 Cell	0.018						
Ni rougher flotation	No. 1 Cell							0.21
	No. 2 Cell		0.028	0.032				
	No. 3 Cell			0.018				
	No. 4 Cell		0.042					
	No. 6 Cell		0.042					
	No. 7 Cell		0.025					
Regrinding	Feed				1.77			
Copper cleaning	No. 1 Cell				0.39			
	No. 2 Cell				0.39			
	No. 4 Cell			0.018				
Nickel cleaning	Thickener	Separan 0.009						
	Condit. Feed					0.35		0.21
	No. 3 Cell		0.035					

	pH	Temp °F
Copper rougher flotation feed	10.3	
Copper precleaner cell No. 1	11.4	72
Copper precleaner cell No. 2	11.4	75
Copper precleaner cell No. 3	11.7	99
Copper precleaner cell No. 4	11.7	98
Copper cleaner feed	12.2	
Copper cleaner cell No. 1	11.5	132
Copper cleaner cell No. 2	11.6	140
Copper cleaner cell No. 3	11.4	142
Copper cleaner cell No. 4	11.5	132

	% Solids
Ball mill discharge	62
No. 1 conditioner feed	40

Screen analysis	
Primary grind (screen u'size)	52.9% -200 mesh
Regrind feed	71.5% -325 mesh
Regrind discharge	92.6% -325 mesh

TABLE 17

Results of Method III Test Run 42

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.84	25.80	0.64	1.0		79.1	4.2
Nickel concentrate	2.88	2.60	6.66	33.30		12.5	68.5
Final tailing	95.28	0.053	0.08		0.29	8.4	27.3
Feed (calcd)	100.00	0.60	0.28			100.0	100.0
Heads (assay)		0.60	0.28				
Copper rougher concentrate		14.80	3.07				
Copper rougher tailing		0.11	0.23				
Copper cleaner tailing		16.60	3.00				
Copper precleaner concentrate		21.20	2.25				
Copper precleaner tailing		9.34	4.00				
Nickel rougher concentrate		1.70	3.07	38.72			
Nickel cleaner tailing		1.16	0.80				

*From Internal Report MS-AC-67-494

In order to confirm results Test Run 42 procedure for copper flotation was repeated in Test Runs 43 and 44. In these 2 test runs, scheme (2) for gangue depression in the nickel cleaners was not employed but was replaced by the addition of sodium silicate to the cleaners. Results are given in Tables 18 and 19.

TABLE 18

Results of Method III Test Run 43

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.59	24.00	1.28	1.56		79.5	8.5
Nickel concentrate	3.00	2.00	5.10	36.34		12.5	63.7
Final tailing	95.41	0.04	0.07		0.24	8.0	27.8
Feed (calcd)	100.00	0.48	0.24			100.0	100.0
Heads (assay)		0.48	0.24				
Copper rougher concentrate		10.40	4.00				
Copper rougher tailing		0.08	0.19				
Copper cleaner tailing		10.80	4.72				
Copper precleaner concentrate		16.00	3.76				
Copper precleaner tailing		6.00	4.38				
Nickel rougher concentrate		1.36	3.22	41.66			
Nickel cleaner tailing		0.76	1.22				

*From Internal Report MS-AC-67-503.

TABLE 19

Results of Method III Test Run 44

Product	Wt %	Assays %*				Distribution %	
		Cu	Ni	Insol	S	Cu	Ni
Copper concentrate	1.49	24.00	1.28	2.30		74.7	8.0
Nickel concentrate	2.49	3.33	5.76	35.84		17.3	59.9
Final tailing	96.02	0.04	0.08		0.23	8.0	32.1
Feed (calcd)	100.00	0.48	0.24			100.0	100.0
Heads (assay)		0.48	0.24				
Copper rougher concentrate		17.34	2.67				
Copper rougher tailing		0.12	0.22				
Copper cleaner tailing		12.88	3.56				
Copper precleaner concentrate		22.00	2.25				
Nickel rougher concentrate		2.00	4.00	40.00			
Nickel cleaner tailing		0.73	1.00				

*From Internal Report MS-AC-67-503

Effect of High Pulp Temperature in Copper Cleaning

In Table 20, the best copper results obtained when copper cleaning was carried out at room temperature (Test Run 40) are compared with results obtained with high temperature cleaning in Test Runs 42, 43 and 44.

TABLE 20

Comparison of Copper Results Obtained Using High and Low Pulp Temperatures in Copper Cleaning

Test Run No.	Temp Range of Pulp in Cleaners °F	Copper Concentrate				
		Wt %	Assays %		Distribution %	
			Cu	Ni	Cu	Ni
40	room temp	1.93	24.00	1.25	79.8	9.3
42	132-142	1.84	25.80	0.64	79.1	4.2
43	108-130	1.59	24.00	1.28	79.5	8.5
44	122-132	1.49	24.00	1.28	74.7	8.0

Retention of Copper Cleaner Tailing as a Separate Product

In the first 4 test runs (No. 35 to 38) the copper cleaner tailing was not recirculated but was saved as a separate product. It was thought that the recirculation of this product in which the nickel minerals had been subjected to regrinding and depression with high lime alkalinity would result in higher nickel losses in the tailing. At the same time it was hoped that enough copper could be floated in the copper cleaning operation to give a copper recovery in the copper concentrate of 80% or higher. The copper cleaner tailing could then be combined with the nickel concentrate. The best set of results using this scheme were obtained in Test Run 36 results of which are given in Table 21.

TABLE 21

Results of Method III Test Run 36

Product	Wt %	Assays %* Cu	Ni	Insol	S	Distribution % Cu	Ni
Copper concentrate	1.45	25.20	1.28	2.60		70.3	7.0
Copper cleaner tailing	2.06	5.40	2.56	35.26		21.4	19.7
Nickel concentrate	1.01	1.44	11.80	24.30		2.8	44.7
Final tailing	95.48	0.03	0.08		0.28	5.5	28.6
Feed (calcd)	100.00	0.52	0.27			100.0	100.0
Heads (assay)		0.52	0.26				
Nickel conc + Cu cleaner tailing	3.07	4.09	5.60	31.65		24.2	64.4

*From Internal Report MS-AC-67-478.

Gangue Depression in Nickel Cleaners

The results obtained using the various schemes for gangue depression in the nickel cleaners are compared in Table 22.

TABLE 22

Comparison of Nickel Concentrate Grades Obtained Using Various Gangue Depression Schemes

Test Run No.	Scheme Used	Product	Wt %	Assays % Cu	Assays % Ni	Assays % Insol	Distribution % Cu	Distribution % Ni
40	Added Na_2CO_3 (0.35 lb/ton) and $CuSO_4$ (0.21 lb/ton) to cleaner feed	Ni cleaner conc	2.17	3.30	6.44	34.16	12.0	53.8
		Ni rougher conc		2.60	4.25	38.24		
42	Thickened before cleaning, then added same reagents as above and also Z-6 (0.035 lb/ton)	Ni cleaner conc	2.88	2.60	6.66	33.30	12.5	68.5
		Ni rougher conc		1.70	3.07	38.72		
44	Added sod. silicate (0.53 lb/ton) to cleaner feed	Ni cleaner conc	2.49	3.33	5.76	35.84	17.3	59.9
		Ni rougher conc		2.00	4.00	40.00		

Distribution of Platinum-Group Metals in Copper and Nickel Concentrates

Samples of concentrates and tailings from Test Runs 42 and 43 were assayed for the platinum group metals and the distribution was calculated. Results are given in Table 23.

TABLE 23

Distribution of Platinum Metals in Copper and Nickel Concentrates

Test Run No.	Product	Wt %	Assays*					Distribution %			
			Cu	Ni	Pt	Pd	Rh	Cu	Ni	Pt	Pd
42	Copper conc	1.84	25.80	0.64	0.076	0.33		79.1	4.2	18.2	22.2
	Nickel conc	2.88	2.60	6.66	0.119	0.575	0.017	12.5	68.5	44.6	60.5
	Final tailing	95.28	0.053	0.08	0.003	0.005		8.4	27.3	37.2	17.3
	Feed (calcd)	100.00	0.60	0.28	0.008	0.027		100.0	100.0	100.0	100.0
43	Copper conc	1.59	24.00	1.28	0.048	0.42		79.5	8.5	12.0	26.5
	Nickel conc	3.00	2.00	5.10	0.092	0.46		12.5	63.7	43.2	54.7
	Final tailing	95.41	0.04	0.07	0.003	0.005		8.0	27.8	44.8	18.8
	Feed (calcd)	100.00	0.48	0.24	0.0064	0.025		100.0	100.0	100.0	100.0

*From Internal Reports MS-AC-67-494, 503 and 568.

CONCLUSIONS AND DISCUSSION

The sulphides in the ore float readily at a coarse grind to produce a marketable grade of bulk concentrate with good recoveries of copper, nickel and precious metals. These results were obtained with the stage addition of small amounts of amyl xanthate and frother. No other reagents were required. The grind was varied from about 50% -200 mesh to 68% -200 mesh with no significant change in results (See Tables 3 and 5).

Although the bulk concentrates were of satisfactory grade, the insoluble content was high - generally 20% or higher. Mineralogical examination showed that this was due to the presence of free gangue minerals, mostly feldspar and orthopyroxene. It was possible by employing various methods and reagents to lower the insoluble content in the concentrate but only at the expense of a drop in recoveries (see Table 3).

There was no change in recoveries when the bulk rougher flotation capacity was increased from 7 to 11 No. 7 Denver cells (compare results of Table 3 with those in Table 5). After passing through 4 rougher cells, tailing losses had not reached their optimum values (see Table 5). This would indicate that the flotation contact time in 7 rougher cells is sufficient to give the best possible recoveries. The calculated contact time for 7 rougher cells is 16.4 minutes (2.2 minutes per cell) for a feed rate of 750 lb per hour, a flotation feed density of 38% solids and taking the specific gravity of the ore as 2.8.

Three stages of cleaning of the bulk rougher concentrate is adequate to upgrade the concentrate to satisfactory values. There was no significant increase in concentrate grade when the number of cleaning stages was increased to five (compare results in Tables 3 and 5).

A comparison of classification by mechanical classifier as against that obtained by a vibrating screen showed that for the same fineness of grind as expressed in terms of -200 mesh, the classifier overflow contained 3% more material on 65 mesh than did the screen undersize (see Table 8). In terms of metal distribution in the various size fractions, the classifier overflow contained about 10% more of both copper and nickel in size fractions below 200 mesh. Because of the differential classification of sulphide and gangue particles due to gravity differences, the product obtained by using a mechanical classifier or a cyclone could be appreciably coarser than the undersize from a vibrating screen for the same degree of sulphide liberation. Therefore it can be assumed that a classified product of about 50% -200 mesh obtained by a vibrating screen could be reduced to 45% -200 mesh or even lower without affecting results by using a mechanical classifier or cyclone in the grinding circuit.

The best selective flotation results were obtained using Method III. Copper concentrate containing approximately 1.3% nickel with 80% copper recovery was consistently made. Although this is a little over the maximum allowable limit of 1.0% nickel, it should be possible with further adjustments in technique to lower the nickel content to the desired specification. In one test run (42) a copper concentrate assaying 0.64% nickel was produced when a high pulp temperature in cleaning was employed but this result was not confirmed in subsequent testing (Test Runs 43 and 44). However, the pulp temperature was not quite as high in these two tests (see Table 19) which may indicate that temperature values are critical. In order to obtain maximum depression of nickel minerals, it was found necessary to regrind the copper rougher concentrate before cleaning and to saturate the pulp in the cleaners with lime.

The nickel concentrates produced by Method III did not meet the required 3 to 1 ratio of nickel to copper content. Generally, this ratio ranged from 2 to 2.5 to 1. However, the specifications for the nickel concentrates are not considered to be as critical as those for the copper concentrate. All the schemes tried in attempts to lower the high gangue content in the nickel concentrate were unsuccessful (see Table 21).

Because smelter payment schedules for the various concentrates are not available it is not possible to compare the net smelter returns for bulk flotation as against selective flotation. However, it would appear that bulk flotation would give the highest return because of the following considerations:

(1) It is possible that there would be no payment for the considerable content of platinum-group metals in the copper concentrate.

(2) No payment for the nickel in the copper concentrate.

(3) Treatment costs for selective flotation would be higher.

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